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MILLING METHODS AND COSTS AT THE BLACK HAWK CONCENTRATOR, HANOVER, NEW MEXICO

BY

IRA L. WRIGHT

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MILLING METHODS AND COSTS AT THE BLACK HAWK CONCENTRATOR, HANOVER, NEW MEXICO[1]

By Ira L. Wright[2]

INTRODUCTION

This paper, describing the milling practice of the Black Hawk Consolidated Mines Co., is one of a series being prepared by the United States Bureau of Mines.

ACKNOWLEDGMENTS

Part of the information contained in this paper was furnished by I. H. Stanley, mill superintendent, Black Hawk Consolidated Mines Co., Hanover, N. M., and a part by the South-western Engineering Corporation, Los Angeles, Calif. The assistance of T. G. Chapman in arranging the paper for publication is also acknowledged.

GENERAL DESCRIPTION

The concentrator of the Black Hawk Consolidated Mines Co. is situated 1 mile south of Hanover, N. M., on the Hanover-Fierro spur of the Santa Fe Railroad. It was designed and built on contract by the Southwestern Engineering Corporation in the summer of 1928. The flow sheet and design were worked out in collaboration with the writer after numerous laboratory tests had been made in various testing plants. The mill was designed to treat 100 tons per day, but certain changes in equipment have increased the capacity to between 160 and 180 tons per day.

A gradual sloping hillside from the Combination shaft, from which most of the ore for the mill is obtained, to the railroad make an ideal mill site and a nearby arroyo provides storage for tailings. The collar of the shaft is 113 feet above the railroad and 800 feet distant.

Water is secured from the Combination shaft, which is 625 feet deep. Up to the present time there has been ample water for two or three times the

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6359."

2 - One of the consulting engineers, U. S. Bureau of Mines, and manager, Black Hawk Consolidated Mines Co.

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capacity of the mill and there has been no indication of a slackening of the supply. As a rule fresh mine water is used rather than used water from the tailings pond. The mine water is comparatively soft and no ingredients detrimental to flotation are found. Power is purchased from the Texas-Louisiana Power Co., which generates it at its Diesel engine plant in Silver City, 16 miles from the mill.

ORE TREATED

The bulk of the ore treated to date and the ore for which the plant was designed as previously stated has come from the Combination shaft situated about 400 feet from the crude ore bin at the crushing plant and connected thereto with an 18-inch gauge tramway. From a small bin at the shaft into which the automatic skip dumps, the ore is trammed by hand to the crude ore bin in a 2-ton capacity car. Most of the ore is mined by overhand stoping, but as the ore bodies are replacements in limestone they are very irregular and the methods of mining are correspondingly irregular. Very little ore is allowed to accumulate in the stopes, so that there is little time for oxidation after the ore is broken.

A small part of the ore treated is from the Lucky Bill mine, 3 miles south of the mill, and is mined by overhand stoping from a vein in porphyry. Some custom ore from various mines in the Central mining district and other districts in the state is also handled. In the last two months about one-half of the ore treated has come from the Ground Hog mine of the Asarco Mining Co. This is high-grade ore and a low ratio of concentration is obtained, about 45 per cent of the crude ore being concentrates.

The average ore milled to date assays about as follows:

Silver	2.0 ounces per ton
Lead	2.5 per cent
Copper	0.5 per cent
Zinc	12.0 per cent

Clean ore will run considerably higher than the average shown, but contamination by limestone and iron pyrite has brought the average down. In recent months the grade of ore has been about 6 per cent lead and 15 per cent zinc, with corresponding silver and copper values.

The metallic minerals are galena, spalerite, chalcopyrite, and pyrite. The association of the silver with the various metallic sulphides is not known. Pyrite is present in amount about equal to the sphalerite. The lead, zinc, and copper minerals are coarsely crystalline and do not require particularly fine grinding to effect a separation.

The gangue consists of limestone, hedenbergite, and garnet. The ore is not hard to crush but part of the gangue is a tough hedenbergite which breaks along radial cleavage surfaces and large thin pieces are common in the ball mill feed. The garnet is not present in sufficient amount to cause excessive wear on the crushers.

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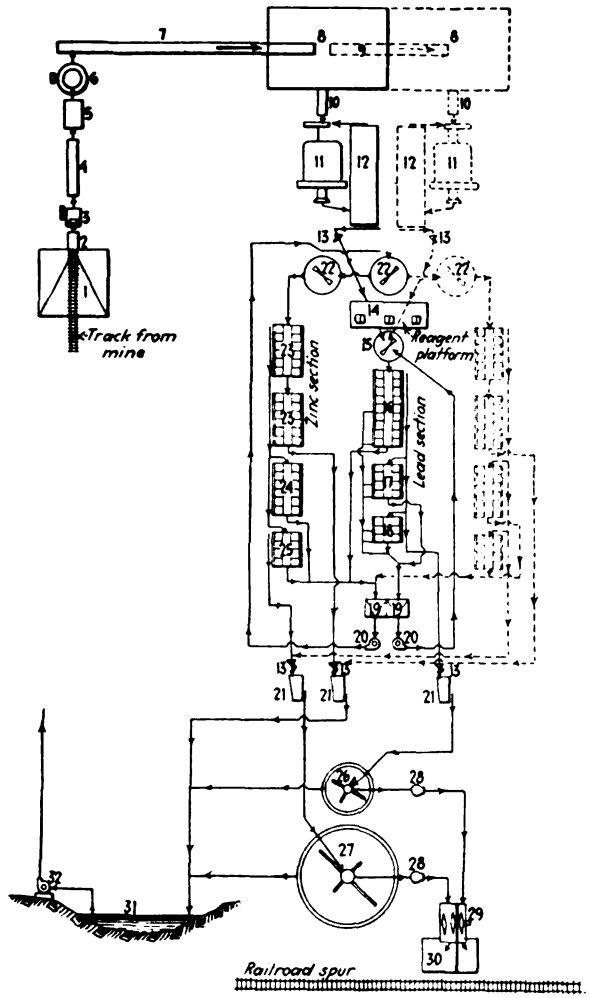

1. Coarse-ore bin.
2. 24-in. apron feeder.
3. 15x24-in. jaw crusher.
4. 16-in. belt conveyor.
5. 30-in. picking belt.
6. 2-ft. Symons cone crusher.
7. 16-in. belt conveyor.
8. Fine-ore bin.
9. 16-in. belt conveyor.
10. 16-in. Southwestern belt feeder.
11. No. 66 Marcy ball mill.
12. 4-ft. 6-in. x 18-ft. Dorr classifier.
13. Southwestern pulp samplers.
14. Southwestern reagent feeders.
15. 5x5-ft. Southwestern conditioner.
16. MB-3012 Southwestern air flotation machine. Lead rougher.
17. MB-3008 Southwestern air flotation machine. Lead cleaner.
18. MB-3006 Southwestern air flotation machine. Lead re-cleaner.
19. Sumps.
20. 2-in. sand pumps.
21. No. 13 Wilfley pilot tables.
22. 6x6-ft. Southwestern conditioners.
23. MB-3012 Southwestern air flotation machines. Zinc rougher.
24. MB-3012 Southwestern air flotation machines. Zinc cleaner.
25. MB-3008 Southwestern air flotation machines. Zinc re-cleaner.
26. 10x8-ft. Dorr thickener.
27. 22x10-ft. Dorr thickener.
28. 2-in. Dorrco suction pumps.
29. 6-ft. diameter, 3-disk American filter.
30. Concentrate bins.
31. Tailing pond.
32. Recovery water pump.

Figure 1.- Flow sheet of the Black Hawk Consolidated mill, in Grant County, N. Mex.

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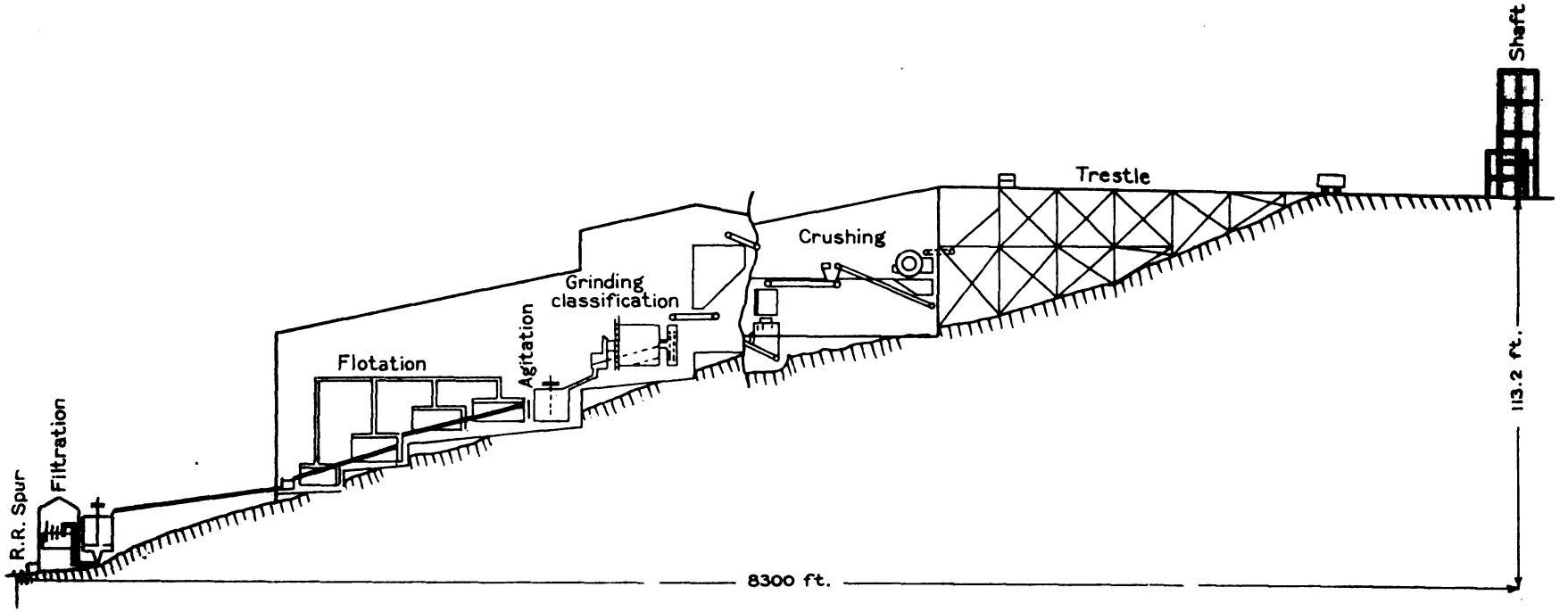

Figure 2.- Diagrammatic section through the combination shaft, crushing plant concentrator and filter building

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The moisture content of ore received by the concentrator varies from 1 to 3 per cent. No change in condition of ore left in stopes has been observed either in appearance or behavior in the mill, except in certain rare instances when an old stope has been started up after a long period of idleness. The first ore broken from such a stope shows some sign of sulphatizing and requires more soda to maintain the usual alkalinity in the flotation circuit.

PRESENT METHOD OF CONCENTRATING

Flow Sheet

Figure 1 shows the flow sheet of the mill as operated at present and a legend indicates the number and sizes of the various machines.

Figure 2 is a diagramatic section through the Combination shaft, crushing plant, mill, and filter building.

The average tonnage concentrated during October, 1929, was 165.5 dry tons per 24 hours, making no allowance for the small amount of lost time. The maximum tonnage handled in 24 hours amounted to 178 dry tons.

Coarse and Intermedate Crushing

The size of material delivered to the coarse crushing plant is governed by material that will pass through chutes in the mine. Pieces of ore larger than 14-inch maximum diameter are rare. Ore from the mine is dumped on a grizzly, with 6-inch openings, located over the crude ore bin of 175 tons capacity. Oversize pieces are broken with sledge hammers. The coarse and intermediate crushing plant operates for eight hours per day and during this period crushes enough material for 24 hours of concentrator operation. Ore is delivered to a 15 by 24 inch Traylor jaw-type crusher from the crude ore bin by a 24-inch apron feeder. The crusher is belt-driven and reduces the ore to a size of 2 1/2 to 3 inches. The crushed product discharges on a 14-inch conveyor belt equipped with a magnetic head pulley for the removal of tramp steel. This conveyor in turn discharges over a small grizzly, with 1-inch openings, to a 36-inch picking belt. The grizzly serves the double purpose of providing a cushion of fine material on which the coarser grizzly oversize drops, thereby reducing wear on the picking belt and placing the cleaned coarser material on top where it can be more readily picked by the ore sorters. Hand-sorting on the picking belt is confined to the removal of waste rock. The removal of waste at this point raises the grade of concentrator heads, permits an increase in mill capacity, reduces wear and tear on machinery, and yields better metallurgical results.

The picking belt discharges into the hopper of a 2-foot Symons cone crusher which reduces the product to minus 1/2 inch size. The crushed product is elevated to a 175-ton capacity storage bin by a 14-inch conveyor belt. At the discharge end of this conveyor belt a Southwestern sampler is provided but is not used unless custom ore is being crushed and sampled.

Table 1 gives a typical screen test of the crushing plant product.

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Table 1.--Typical screen test of crushing plant product

Screen size	Per cent	Cumulative per cent
On 1/4 inch	4.28	4.28
Through 1/4 inch on 1/8 inch	29.74	34.02
Through 1/8 inch on 8 mesh	16.65	50.67
Through 8 mesh on 20 mesh	18.55	69.52
Through 20 mesh on 40 mesh	11.33	80.55
Through 40 mesh on 60 mesh	5.14	85.69
Through 60 mesh on 80 mesh	2.70	88.39
Through 80 mesh on 100 mesh	1.83	90.22
Through 100 mesh on 200 mesh	3.82	94.04
Through 200 mesh	5.95	99.99

Grinding

An 18-inch belt-feeder operated by an adjustable ratchet mechanism feeds the ore from the fine-ore storage bin to one 6 by 6 foot Marcey ball mill. The pinion shaft of the ball mill is directly connected through Timken bearing, Falk flexible coupling, and Culter-Hammer magnetic clutch to a 125-horsepower General Electric synchronous motor which operates the ball mill at 25 revolutions per minute. This motor is larger than necessary but was selected, rather than a smaller one, in order to obtain a better power factor for the whole mill. The ball load ranges from 12,000 to 14,000 pounds and replacements are of 4-inch chrome steel balls. The ball consumption has averaged 1.5 pounds per ton of ore ground. Manganese steel liners are used, and the first set installed gave a life equivalent to 24,050 tons of ground ore.

The ball mill operated in closed circuit with a 6 by 18 foot Dorr duplex classifier. There is considerable variation in the amount of circulating load, but this load does not often exceed three times the feed tonnage. The pulp density in the ball mill averages 72 per cent of solids and that of the classifier overflow varies from 30 to 34 per cent of solids.

Table 5 gives a typical screen test of the classifier overflow product.

The original Dorr classifier installed was Model "C", 4 1/2 by 18 foot size. From the start of concentrator operations in August, 1928, trouble was experienced in the grinding circuit due to the overloading of this classifier. It was found that with this classifier in service a maximum tonnage of but 115 tons could be obtained in the grinding circuit. This machine was replaced with a Model "D" duplex classifier of 6 by 18 foot size and this change resulted in an increase of maximum tonnage ground to 180 tons per 24 hours, without sacrifice in fineness of grinding. The following figures show reductions in milling costs due to the operation of the larger classifier.

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Year	Month	Operating cost per ton
1928	September	$ 1.95 with small classifier
1928	October	$ 1.86 with small classifier
1928	November	$ 2.04 with small classifier
1929	September	$ 1.62 with larger classifier
1929	October	$ 1.50 with larger classifier
1929	November	$ 1.65 with larger classifier

Flotation

Figure 3 gives the flotation flow sheet in detail and the legend accompanying this figure gives the sizes of machines. All flotation machines used are Southwestern air-lift-type cells.

The classifier overflow product goes to a 4 by 5 foot conditioning tank and from there to a double-spitz 12-foot by 30-inch flotation machine operating as a lead rougher. The concentrates from the first section of the lead rougher are cleaned in one double-spitz 8-foot by 30-inch cleaner cell, and the concentrates of the cleaner cell are again cleaned in one 6-foot by 30-inch recleaner cell, the recleaner cell producing finished lead concentrates which go to the lead concentrates thickener. The concentrates overflowing the second section of the lead rougher, together with the middlings product of the cleaner and recleaner cells, are returned to the conditioning tank ahead of the lead rougher cells by a 2-inch Wilfley sand pump.

Figure 3.--Flotation flow sheet legend

1. Conditioner, 4 by 5 foot
2. Lead rougher, double spitz, 12 foot by 30 inch
3. Lead cleaner, double spitz, 8 foot by 30 inch
4. Lead recleaner, double spitz, 6 foot by 30 inch
5. Wilfley samd pump, 2 inch
6. Conditioner, 6 by 6 foot
7. Conditioner, 6 by 6 foot
8. Zinc rougher, double spitz, 12 foot by 30 inch
9. Zinc rougher, double spitz, 12 foot by 30 inch
10. Zinc cleaner, double spitz, 12 foot by 30 inch
11. Zinc recleaner, double spitz, 8 foot by 30 inch
12. Wilfley sand pump, 2 inch

The tailings product of the lead rougher cells is elevated to two 6 by 6 foot conditioning tanks by a 2-inch Wilfley sand pump. The conditioning tanks operate in series and deliver feed to two double-spitz 12-foot by 30-inch flotation cells operating in series as zinc roughers. The concentrates from the rougher and the concentrates from the first section of the second rougher are cleaned in one 12-foot by 30-inch flotation cell, and the concentrates of the cleaner cell are again cleaned in one 8-foot by 30-inch recleaner cell. The

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recleaner cell concentrates are a finished product and go to the zinc concentrates thickener. The tailings product of the second rougher cell consists of t final mill tailings and goes to the tailings settling pond. The middlings produ of the cleaner and recleaner zinc cells is returned to the conditioning tanks at the head of the zinc section by the 2-inch Wilfley sand pump already mentioned.

The kind and amount of flotation reagents are changed within wide limits due to the variation in the ore from the Combination mine and also due to the variation in custom ores received at the concentrator.

Table 2 gives a list of the reagents employed, an approximate average of the amount of each used, and the place of adding each reagent. The reagents listed have not all been used at the same time.

Table 2.--Flotation reagents

Reagent	Pounds added per ton of ore	Added to
Thiocarbanalid	0.05	Ball mill feed
Soda ash	0.80	Ball mill feed
Pine oil	0.07	Classifier overflow
Cresylic acid	0.05	Classifier overflow
Sodium ethyl xanthate	0.10	Lead rougher
Sodium cyanide	0.30	Lead cleaner
Sodium cyanide	0.10	Lead recleaner
Zinc sulphate	0.50	Lead cleaner
Copper sulphate	1.40	Lead section tailing launder ahead of pump
Sodium ethyl xanthate	0.20	Zinc conditioner
Barrett No. 634 creosote	0.05	Lead section tailings launder
Lime (fed dry)	0.20	Zinc cleaner

Southwestern reagent feeders are used throughout the mill. Soda ash, lime, and thiocarbanalid are fed dry. For a time the use of lime as a depressant for pyrite in the zinc recleaner was discontinued as it was believed that lime in the return water was detrimental to the flotation of galena. Later experience seemed to disprove this theory, especially when thiocarbanalid is used instead of xanthate as a collector in the lead circuit.

At the beginning of operations at this plant, it was considered best to add the soda ash, the sodium cyanide and the zinc sulphate to the ball mill, but subsequently the sodium cyanide and the zinc sulphate were added to the lead rougher concentrates ahead of the cleaner cell and additional amounts of these were added to the cleaner lead concentrates before reaching the recleaner cell. This change resulted in a more concentrated solution of these reagents in their roll as zinc depressants and produced a higher grade of lead concentrates. Since the middlings from the lead cleaner and recleaner cells are returned to the

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lead rougher cell certain portions of these two reagents are present in the lead rougher to act as zinc depressants.

Provision has recently been made to by-pass a portion of the concentrates from the lead rougher, cleaner and recleaner cells to a pump sump and return these products to the lead rougher cell. Adjustable cutters are provided so that a small or large portion of the concentrates, of any one of the machines mentioned, may be returned. The return of these products to the lead rougher cell has allowed a better regulation of grade of concentrates and has produced a more uniform condition in the lead flotation circuit.

Air is furnished to the flotation cells by a 24-inch Connersville blower operating at 350 revolutions per minute and requiring 60 horsepower. The blower delivers approximately 9,000 cubic feet of free air per minute at a pressure of 1 1/3 pounds.

The pulp in the lead rougher contains from 30 to 35 per cent of solids and the pulp of the zinc roughers contains about 20 per cent of solids. Pulps are lifted from the lower end of the flotation sections to the conditioners, a lift of 20 feet, by 2-inch Wilfley sand pumps. The greater load is on the pump which lifts the tailing from the lead rougher cell, together with middlings from the zinc cleaner and recleaner cells, to the conditioners at the head of the zinc section. This pump is not taxed to the limit of its capacity, although at times 40 tons of pulp are handled per hour.

At the time the concentrator was built there seemed to be no record of experience, by flotation operators, in the use of the air-lift type of flotation machine for both the lead and zinc sections of a mill using selective flotation methods. Some misgivings as to the suitability of the air-lift type of machine for selective work was felt, especially in the lead section, and many operators were of the firm opinion that some type of mechanically agitated flotation machine would be preferable for the lead section. It is not possible, after one and a half years of experience, to state which type of flotation machine is best for ore treated at this plant since mechanically agitated machines have not been tried. However, experience with this concentrator when using the same ore that is being treated in another concentrator by mechanically agitated flotation machines, indicates that the metallurgical results obtained with the air-lift machines are as good as with the mechanically agitated machines. Experience at this plant indicates that the successful treatment of lead-zinc ores by selective flotation methods depends upon the proper conditioning of the pulp with the correct reagents rather than upon the type of flotation machine used.

No attempt has been made to segregate primary slime for separate flotation treatment. Screen analyses of flotation tailings show that the greatest loss occurs in the minus-200-mesh size material, but it is not known whether this loss is due to primary slime or to the fact that all slime is less susceptible to the actions of the various reagents than the more granular material.

Table 3 gives average analyses of lead and zinc concentrates produced in September and October, 1929.

Table 4 gives typical analyses of values in the concentrates produced

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and also the recoveries of values for September 5, 1929.

Table 5 shows the distribution of copper, lead, and zinc in typical concentrator heads and Table 6 shows the distribution of the same metals in tailings, the samples represented in Tables 5 and 6 being taken for the same period.

Table 3.--Average analyses of lead and zinc concentrates for September and October, 1929

Product	Per cent								Ounces per ton	
	Lead	Copper	Zinc	Sulphur	Insoluble	Iron	Bismuth	CaO	MgO	Silver	Gold
Lead concentrates	59.30	3.68	11.37	19.43	1.15	--	0.138	---	---	25.22	--
Zinc concentrates	1.81	1.55	54.77	---	1.61	5.50	---	0.82	Tr.	1.78	Tr.

Table 4.--Typical analyses and metal recoveries in lead and zinc concentrates

Product	Weight, tons	Analyses				Per cent of total metal recovered or lost			
		Silver, ounces per ton	Lead, per cent	Copper, per cent	Zinc, per cent	Silver	Lead	Copper	Zinc
Heads	145.90	1.44	1.97	0.57	11.0	--	--	--	--
Lead concentrates	4.65	29.86	52.50	5.38	13.1	66.15	84.90	30.11	3.60
Zinc concentrates	26.13	2.04	1.32	1.87	56.0	25.37	11.09	58.76	91.18
Tailings	115.12	0.12	0.10	0.08	0.7	6.58	4.01	11.07	5.02

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Table 5.-- Distribution of copper, lead, and zinc in typical concentrator heads

Product, mesh	Weight, per cent	Analyses, per cent			Per cent of total metal		
		Copper	Lead	Zinc	Copper	Lead	Zinc
On 100 mesh	7	0.14	0.32	2.4	2.1	0.9	1.4
Through 100 on 200 mesh	18	0.31	0.64	10.8	12.2	4.8	1.6
Through 200 mesh	75	0.52	3.02	15.4	85.7	94.3	97.0
Composite	100	0.46	2.40	11.9	100.0	100.0	100.0

Table 6.--Distribution of copper, lead, and zinc in typical concentrator tailings

Product, mesh	Weight, per cent	Analyses, per cent			Per cent of total metal		
		Copper	Lead	Zinc	Copper	Lead	Zinc
On 100 mesh	12	0.12	0.18	1.4	9.5	6.2	7.8
Through 100 on 200 mesh	21	0.14	0.28	1.5	19.5	16.8	14.6
Through 200 mesh	67	0.16	0.40	2.5	71.0	77.0	77.6
Composite	100	0.15	0.35	2.2	100.0	100.0	100.0

DEWATERING AND HANDLING OF CONCENTRATES

Both lead and zinc concentrates flow to their respective thickeners, a distance of 300 feet, in 6-inch open launders. At present the lead concentrate pulp is thickened in a 10 by 8 foot Dorr thickener which has sufficient capacity when the concentrator is operating on Combination ore. Due to the fact that in the near future nearly the entire tonnage to be treated is expected to be custom ore, which contains more recoverable lead and copper minerals than Combination ore, a larger thickener, 22 by 10 foot in size, is being installed to replace the lead concentrates pulp thickener now in service. The zinc concentrates pulp is thickened in a 22 by 10 foot Dorr thickener. The concentrates pulp fed to both thickeners contain less than 5 per cent of solids, and the thickener discharges contain about 50 per cent of solids. The alkalinity of the lead concentrates pulp corresponds to a pH value of about eight and that of the zinc concentrates pulp to about ten.

The thickened pulps are fed to a 6-foot, three leaf, American filter, two leaves filtering zinc pulp and the remaining leaf filtering lead pulp. A 12 by 6 inch Ingersoll-Rand compressor, type ER-1, produces a vacuum of between 16 and 20 inches of mercury and the blowing air is furnished by a No. 12 Connersville belt-driven blower. Filter bags are purchased from the Filter Fabrics Co. of Salt Lake City. The average life of a bag is four months, which is equivalent to filtering 60 tons of dry concentrates per square foot, if the filter operates continuously at full capacity. The filter has handled up to 60 tons of dry concentrates in 24

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hours and could handle more if the concentrates were available. It is likely that 75 tons of dry concentrates could be filtered, producing a cake containing 10 per cent of moisture, this capacity being equivalent to 1000 pounds of dry concentrates per square foot of filter surface per 24 hours.

The lead concentrates cake averages about 10 per cent of moisture and less than 1 per cent of insoluble, and the zinc concentrates cake averages about 11 per cent of moisture and 1 per cent of insoluble. The smelter moisture is from 1 to 3 per cent less than that obtained at the concentrator, the decrease being due to absorption of moisture in the dry cars. Lead and zinc concentrates drop from the filter into separate bins from which they are loaded into box cars by wheelbarrows. The weight, per cubic foot of concentrates, is used in gaging the amount of concentrates to put into a car, and results show that the cars are rarely overloaded or underloaded. All water from filtering operations flows into a tank from which it is pumped to the return-water storage tank at the head of the concentrator.

DEWATERING AND DISPOSAL OF TAILINGS

The concentrator tailings are elevated, by means of a 2-inch Wilfley pump, to a sufficient height to allow flow by gravity to the launder on the tailings dam. Holes in the bottom of this launder, at regular intervals, allow the tailings to be drawn off where needed to keep the dam building up evenly. A bank of dried coarse tailings 2 to 3 feet high is maintained about 5 feet outside the launder by hand-shovelling to prevent water and slime from running over and starting a break. The tailing pulp is not thickened before impounding, and only such water as becomes clear by settling is drawn off for reuse. About 60 per cent of the total water in the tailings pulp is clarified by settling, but all of this is not returned to circulation, as it is not needed. The tailings pond must be partly drained, at intervals allowing excess water to flow to waste.

At the start of operations some trouble was encountered from breaks in the tailings dam. At one time the center section slid and had to be built up and reinforced with bags filled with sand, old barrels filled with sand, and some loose earth. Since the dam has increased in size and each section has time to dry out between intervals of building up, no trouble with slides has been experienced. As only clear water is ever allowed to flow into the main drainage creek, no water pollution difficulties have been encountered.

SAMPLING

Sampling of concentrator heads overflowing the drag classifier, lead concentrates, zinc concentrates, and final tailings is done with Southwestern samplers. These samplers consist of a channel attached to an arm which is moved by a piston to draw the channel across the stream at regular intervals. The piston is operated with compressed air released by a central timing mechanism. At first water was used in the system, but freezing temperatures prevented the satisfactory operation of those samplers situated outside of the mill building.

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WATER SUPPLY

The fresh water supply is obtained entirely from the Combination mine and contains no impurities harmful to the flotation process. The amount pumped from the mine per 24 hours ranges between 80,000 and 100,000 gallons. The fresh-water storage tank, receiving the mine water, has a capacity of 20,000 gallons. The concentrator-water storage tank has a capacity of 10,000 gallons and receives water returned from the tailings pond and the balance from the 20,000 gallon capacity fresh water tank. The fresh water consumption per 100 tons of ore milled amounts to about 40,000 gallons. Less fresh water and more return water from the settling pond could be used without detriment to flotation operations if economy of water was necessary. Clarified water from the tailings pond, from the concentrate thickeners and from the filter is collected in a common sump tank from which it is pumped to the concentrator water storage tank by a 6 by 8 inch Gould triplex pump. Difficulties in flotation operations from accumulated reagents in the return water have not been experienced.

METALLURGICAL CONTROL

For control of flotation conditions the operators depend on visual inspection of the concentrate froths, on the appearance of products fed to Wilfley laboratory pilot tables and on the panning of various products. Small portions of the final zinc concentrates and of the final tailings are passed over Wilfley laboratory pilot tables. The quality of the zinc concentrates, with respect to lead, pyrite, and zinc contents, is clearly indicated and operators are able to estimate the zinc content, at any time, within 1 or 2 per cent. The pilot table, treating the final tailings product, indicates the amount of zinc not floated and also, on rare occasions, a lead streak. The appearance of a lead streak on the pilot table indicates that the operation of the lead flotation section is very poor. The quality of lead concentrates is judged by panning a sample of the lead recleaner froth.

The experienced operator is able to tell a great deal about conditions by the physical appearance of the froth and changes certain reagents which experience leads him to believe will correct undesirable conditions. At the beginning of the shift, the operator in charge inspects the froths on the various flotation machines and on the pilot tables and pans the lead concentrate. He next checks up on the amount of each reagent used by the operator on the previous shift. It is left to the operator on charge of the shift to control reagents. No attempt is made to dictate what shall be done in any specific case, as it has not been found that chemical analyses of the water or the ore being treated will indicate the amount or kind of reagents to use.

During the shift the operator is required to note in the shift record book the kind and amount of each reagent added as well as the place of adding each reagent. Hourly records are made of the measured tonnage of ore entering the ball mill and the dilution of the classifier overflow product. The amount of ore entering the ball mill which gives the rate of millins, is determined by drawing and weighing the full stream of ore for one minute. The dilution of classifier overflow is determined by weighing an exact gallon of this product. The percentage

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of solids corresponding to the weight of a gallon of pulp is found from a table.

At the close of the shift, the operator notes any unusual conditions encountered and gives a summary of the kind of shift he has had. Assays of samples taken on an operators shift are furnished to him as soon as obtained, which is usually 24 hours after samples are taken. The tailings product of the lead section which is the feed to the zinc section is not sampled.

Metallurgical results are calculated by means of the usual three product-recovery formulas. The average assays of the three shifts of each day are taken for the calculations. Only lead and zinc assays are used to obtain lead and zinc recoveries and ratios of concentration. Copper and silver recoveries are computed by applying assays in the two metal formulas to the various percentages of material in the lead concentrates, zinc concentrates, and final tailings. The account of these two metals seldom adds exactly to 100 per cent.

The amounts of ore in the coarse and in the fine ore storage bins are extimated each morning at seven o'clock. These estimates, together with the weight of ore put into the coarse ore storage bin during the 24 hours, give information for computing the tons of ore milled in the previous 24-hour period. The metallurgical results applied to tons milled give the tons of each of the concentrates produced. Returns from the smelter are less than the calculated amount by about 3 per cent. A gun sample of each carload of concentrates is taken for moisture and metallic contents before the car leaves the siding. These samples have checked smelter samples very closely.

In addition to metallurgical recoveries computed as described, a "commercial recovery" is found by dividing the value of the concentrates actually produced by the value that would be obtained if perfect recovery and separation had been made, producing "perfect" concentrates. The analyses of the "perfect" concentrates are assumed from a consideration of the mineralogical composition of the ore. "Perfect" zinc concentrates are usually assumed to contain 56 per cent of zinc. The composition of "perfect" lead concentrate will vary with the copper content of the ore, the copper usually occuring as chalcopyrite. When the concentrator is operating entirely on Combination ore "perfect" lead concentrates are assumed to contain 50 per cent of lead. The ratio of concentration, for perfect lead concentrates and perfect recovery, would be obtained by dividing the assumed percentage of lead in "perfect" lead concentrates by the per cent of lead in the heads to the concentrator. The assay values of the silver and copper in the heads multiplied by the ratio of concentration gives the assay values of the "perfect" concentrates for these two metals. The assumed analysis of each of these "perfect" concentrates remains the same, unless an ore of radically different composition is delivered to the concentrator as occurred recently when a large proportion of the mill feed was custom ore from a neighboring mine.

The values of the "perfect" concentrates and the concentrates actually produced are obtained by applying the current metal quotation to the smelter contracts in force and deducting freight and treatment charges. Table 7 shows metallurgical results and Table 8 shows the commercial recovery figured for

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Table 7.--Metallurgical results for February 27, 1930.

Product	Weight: tons	Analyses: Silver: ounces per ton	Analyses: Lead: per cent	Analyses: Copper: per cent	Analyses: Zinc: per cent	Recoveries, per cent: Silver	Recoveries, per cent: Lead	Recoveries, per cent: Copper	Recoveries, per cent: Zinc	Value per ton: dollars	Total value of production: dollars
Heads	125.8	5.84	6.5	2.51	14.2	--	--	--	--	---	----
Lead concentrates	[1] 20.70	31.84	36.7	13.65	14.0	89.74	92.94	89.51	16.23	65.16	1,348.81
Zinc concentrates	[1] 24.47	2.36	1.7	1.18	52.9	7.86	5.09	9.14	72.49	14.02	343.07
Tailings	80.63	0.48	0.2	0.05	2.5	5.27	1.97	1.35	11.28	---	----
Total	---	---	--	--	--	102.87	100.--	100.00	100.--	---	1,691.88

[1] Calculated

Table 8.--"Commercial" recovery computed for February 27, 1930

Product	Maximum possible weight, tons	Analyses: Silver, ounces per ton	Analyses: Lead per cent	Analyses: Copper per cent	Analyses: Zinc per cent	Value per ton: dollars	Total value: Dollars	"Commercial" recovery, per cent
"Perfect" lead concentrate	20.90	35.16	40.0	15.11	--	[1] 74.59	1558.93	-----
"Perfect" zinc concentrate	31.93	---	---	---	56.0	[1] 16.49	526.52	-----
Total	---	--	--	--	--	---	2085.45	$\frac{1,691.88}{2,085.45}$ = 81.1

[1] Metal prices used are as follows:
Silver - 43.375 cents per ounce
Lead - 6.1 cents per pound
Copper - 17.75 cents per pound
Zinc - 5.10 cents per pound

February 27, 1930. Table 9 gives the calculated metallurgical report averaged for September and October, 1929. Table 10 contains summaries of total milling costs and milling costs per ton of ore concentrated averaged for September and October, 1929, and Table 11 gives milling costs in terms of reagents, power, labor, assaying, and miscellaneous supplies averaged for the same period.

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Table 9.--Metallurgical report, average of September and October, 1929

	Weight, tons	Analyses: Per cent Pb	Cu	Zn	Insoluble	Fe	Mn	CaO	MgO	S	Bi	Ounces per ton: Ag	Au	Recoveries, per cent: Silver	Lead	Copper	Zinc
Heads	9,221.6	1.99	0.43	9.93	---	---	---	---	---	---	---	1.155	---	---	---	---	---
Lead concentrates	255.2	58.37	3.68	11.57	1.15	---	---	---	---	19.43	0.138	25.22	---	62.95	81.86	23.45	---
Zinc concentrates	1,465.8	1.81	1.55	54.77	1.61	5.5	.67	.82	Tr	---	---	1.78	Tr	---	---	---	90.69

Table 10.--Summary of costs averaged for September and October, 1929

	Total cost	Cost per ton
Crushing	$ 1,017.60	$ 0.1110
Sorting	436.13	.0472
Grinding	3,276.82	.3553
Flotation	4,852.72	.5262
Thickening and filtering	754.21	.0817
General	2,256.95	.2447
Assaying	1,027.32	.1114
Miscellaneous	674.60	.0731
Totals	$ 14,296.35	$ 1.5506

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Table 11.--Summary of costs averaged for September and October, 1929

	Total cost	Cost per ton
Reagents	$ 2,783.72	$ 0.3018
Power	3,176.85	.3445
Labor	4,701.93	.5098
Miscellaneous	3,633.79	.3945
Total	$ 14,296.35	$ 1.5506

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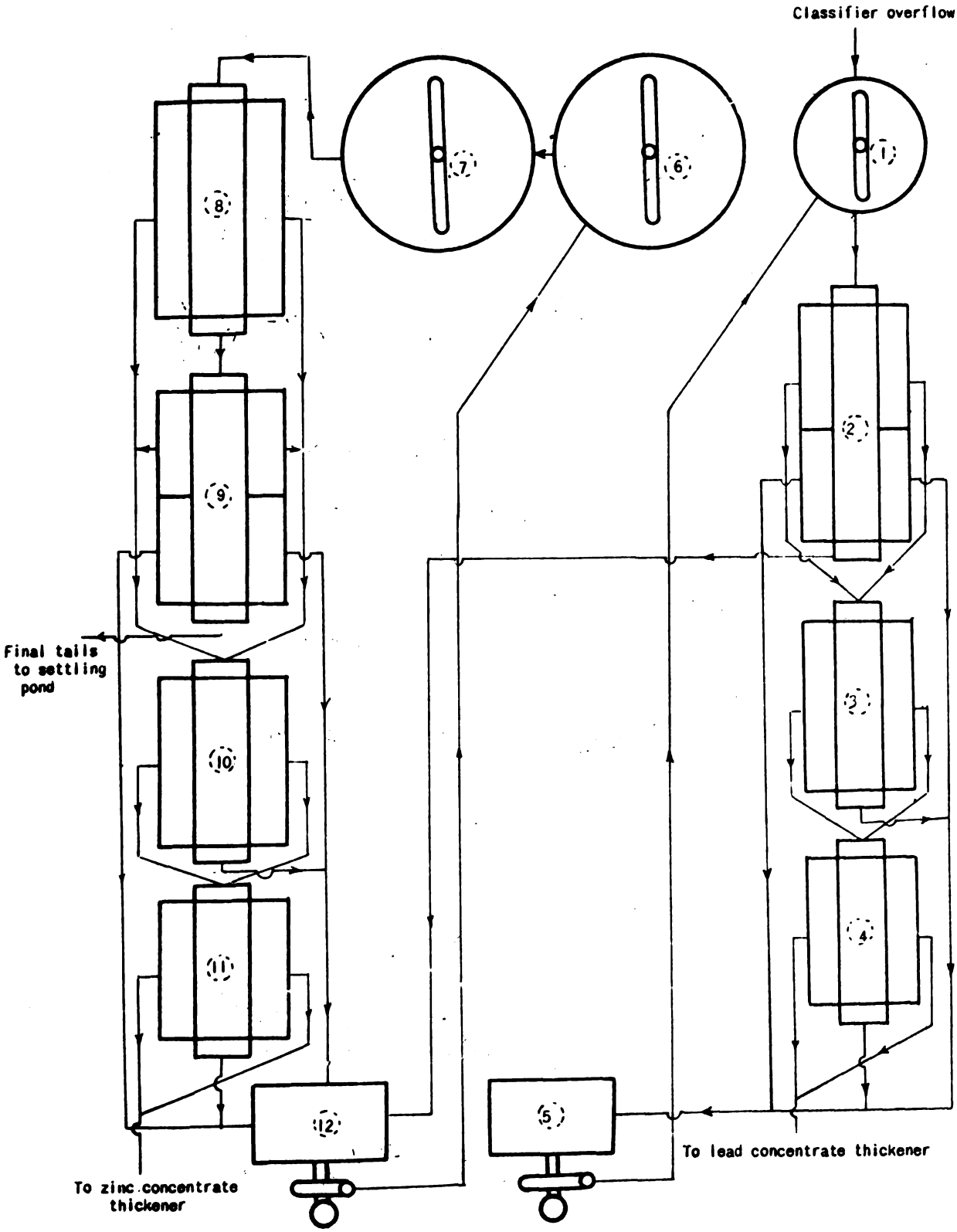

Figure 3.- Flotation flow sheet

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